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MINING PRACTICE AT THE PECOS MINE OF THE AMERICAN METAL CO. OF NEW MEXICO

BY

J. T. MATSON AND C. HOAG

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MINING PRACTICE AT THE PECOS MINE OF THE AMERICAN METAL CO. OF NEW MEXICO[1]

By J. T. Matson[2] and C. Hoag[3]

INTRODUCTION

This paper describing the mining practice at the Pecos mine of the American Metal Co. of New Mexico, a subsidiary of the American Metal Co., Ltd., is one of a series being prepared by the Bureau of Mines on mining methods, practices, and costs in the various mining districts of the United States.

The mine is producing about 18,000 tons per month of zinc-lead-copper ore, chiefly by the square-set and cut-and-fill methods. Costs are given for 1927, 1928, and 1929, the whole producing life of the mine to date.

The Pecos mine is situated at Tererro, San Miguel County, northern New Mexico, in the narrow valley of the upper Pecos River at its junction with Willow Creek, as shown in Figure 1. It is 17 miles northeast of Santa Fe, capital of the State, and 14 miles north of Glorieta, the nearest station on the main line of the Atchison, Topeka and Santa Fe Railway. A standard gage spur 4 1/2 miles long connects the main line to the mill at Alamitos, and a 12-mile aerial tramway connects the mill to the mine. The mine is also served by a good automobile road from Pecos, on U. S. Highway 85 (the old Santa Fe Trail) 16 miles down the river. The altitude at the mine is 8,000 feet.

The authors acknowledge the assistance of E. C. Anderson, chief engineer, and J. H. Russell, chief accountant, of the Pecos mine, and of E. D. Gardner, mining engineer, and C. H. Johnson, assistant mining engineer, of the U. S. Bureau of Mines.

HISTORY

The Pecos mine is the only producing metal mine in the district. It was first located in the early eighties. Development was carried on intermittently until the mine was taken over by the Goodrich-Lockhart Co. in 1916. From 1916 to 1925 this company did 10,000 feet of development work and 17,000 feet of diamond drilling. The American Metal Co., Ltd., acquired the property in 1925.

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[1] - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6368."

[2] - General manager, Pecos mine, and one of the consulting engineers, U. S. Bureau of Mines.

[3] - Mine superintendent, Pecos mine, and one of the consulting engineers, U. S. Bureau of Mines.

Prior to this time no ore had been produced, except such material as was taken out in the course of development work and left on the dump. The unproductiveness of the mine was due in large part to the lack of a profitable method of treating the complex ore.

From early in April, 1926, to the end of that year, an intensive construction and development program was carried out to prepare the mine for production. This included the construction of a 600-ton selective flotation mill, a power plant, two townsites, a 12-mile aerial tramway, a crushing plant, and the mine surface plant. A new shaft was sunk to the 400 level; this level was equipped for the main haulage way, and an upper level known as the waste level was prepared to supply waste for all mining above the 400. The principal part of this work was finished before 1927, at a cost of approximately $2,000,000. The mill was started in January, 1927. From that time to January, 1930, the mine has produced 584,158 tons of ore averaging 16.06 per cent zinc, 3.73 per cent lead, 1.02 per cent copper, 3.39 ounces of silver, and .109 ounce of gold per ton.

The lead concentrates are shipped to the smelter of the American Smelting and Refining Co. at El Paso, Tex. The zinc concentrates are shipped to the Blackwell zinc Co., Blackwell, Okla., and to the American Zinc and Chemical Co., at Langeloth, Pa.

GEOLOGY

The Pecos mine is in the heart of the main mass of the pre-Cambrian rocks of New Mexico. In the neighborhood of the mine, however, the granite is exposed only in the canyons of the Pecos River and its tributary creeks; elsewhere it is covered by extensive areas of the Magdalena limestone formation of Carboniferous age. The pre-Cambrian granite has been intruded by a dark, fine-grained diroite. An extensive northeast-southwest shear zone has altered the igneous rocks into parallel bands of schist. This zone, which is from a few feet to several hundred feet wide, contains rocks of all degrees of metamorphism and varying from acid to basic. The ore bodies are found in the shear zone, replacing the schist. In only a few places have they been found to reach up to the bottom of the limestone, and in no case to extend into it.

The mineralization consists of a mixture of sphalerite, galena, chalcopyrite, and pyrite, carrying appreciable values in gold and silver, associated with the products of metamorphism and alteration, such as talc, hornblende, mica, and chlorite schist. Near the surface the ore bodies have been oxidized, but to only shallow depths. No secondary enrichment is observable. In the Evangeline ore body, lead appears to increase in proportion to zinc as depth is gained.

PHYSICAL CHARACTERISTICS OF THE ORE AND WALL ROCKS

The ore bodies have been developed for a length of 2,000 feet and from the surface to the 1200 level. They consist of irregular, disconnected, often overlapping lenses of sulphides in the shear zones. The lenses often pinch out abruptly, either horizontally or vertically. Figures 2-a and 2-b illustrate the development of the two main ore bodies, the Katydid and the Evangeline, and some

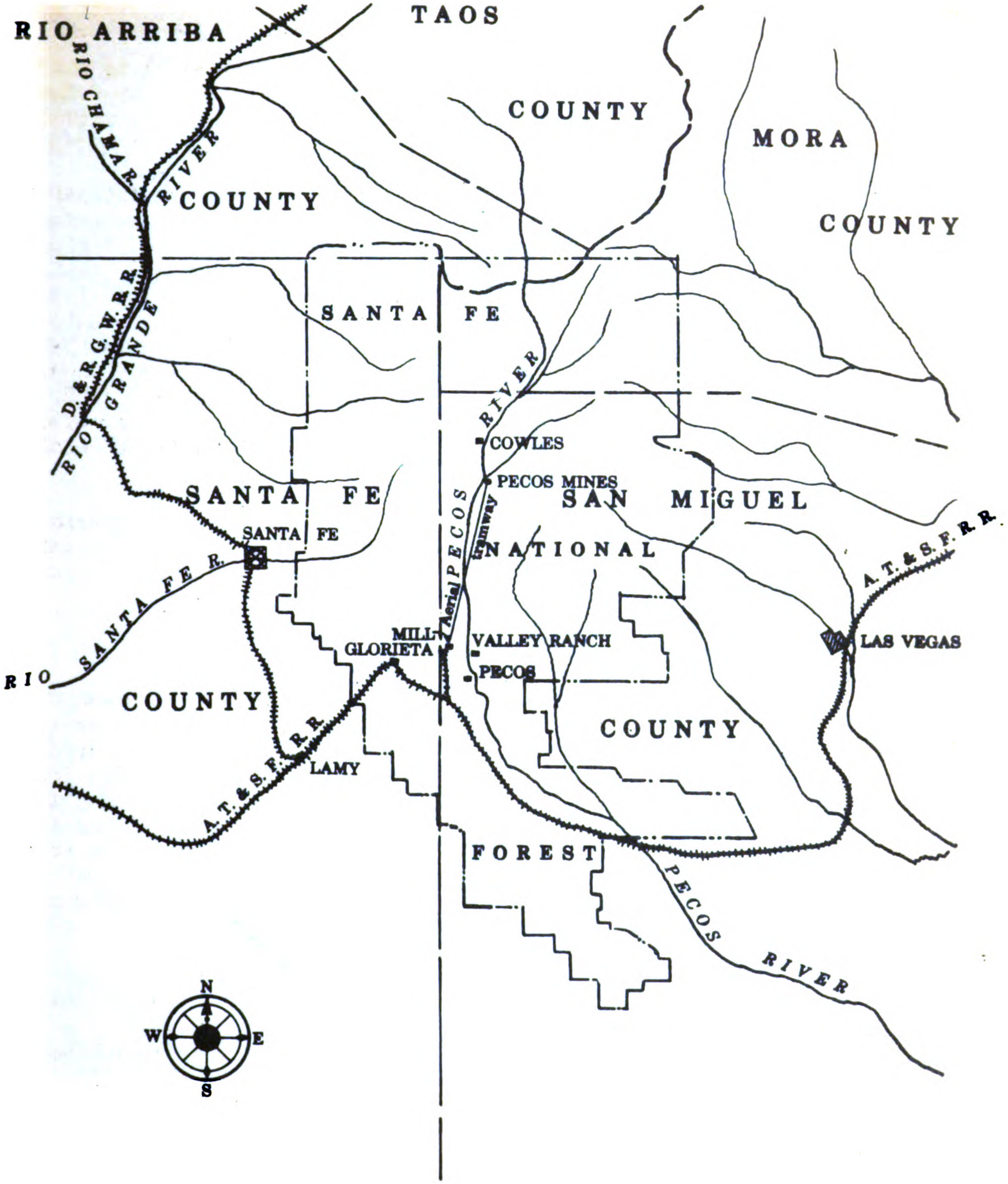


Figure 1.—Location of the Pecos mines, New Mexico

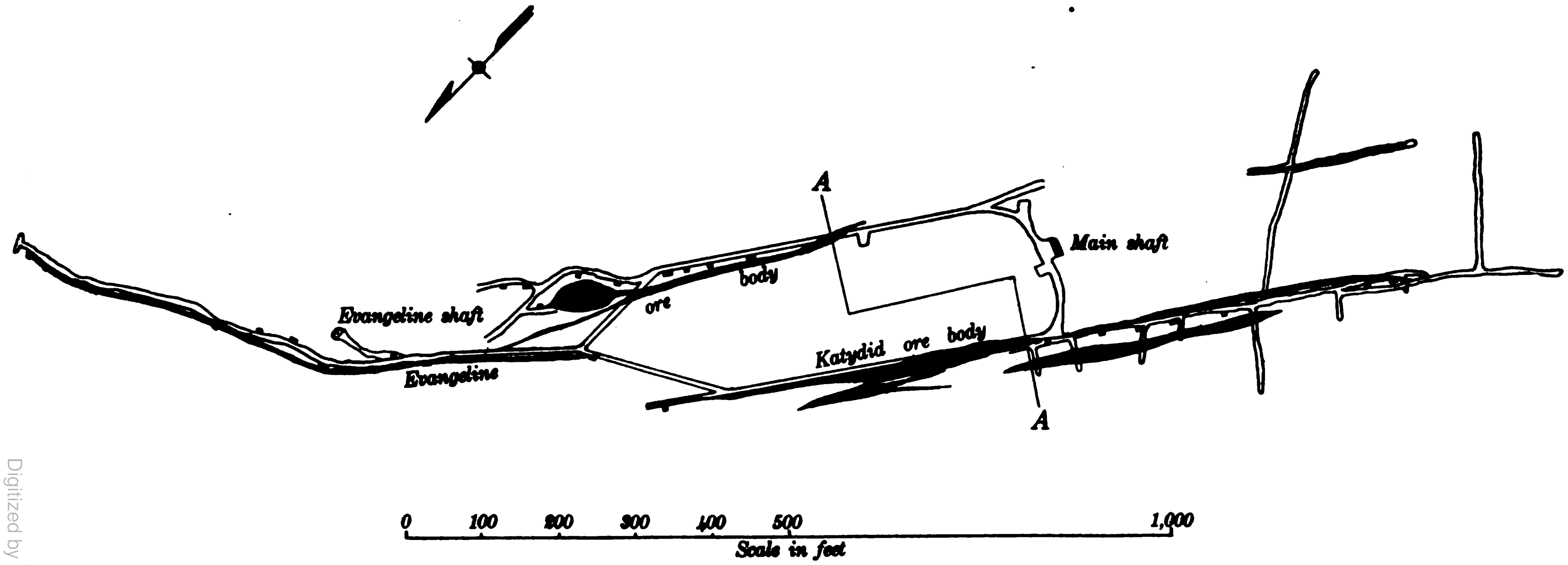


Figure 2a.--Plan of 400 level, Pecos mines

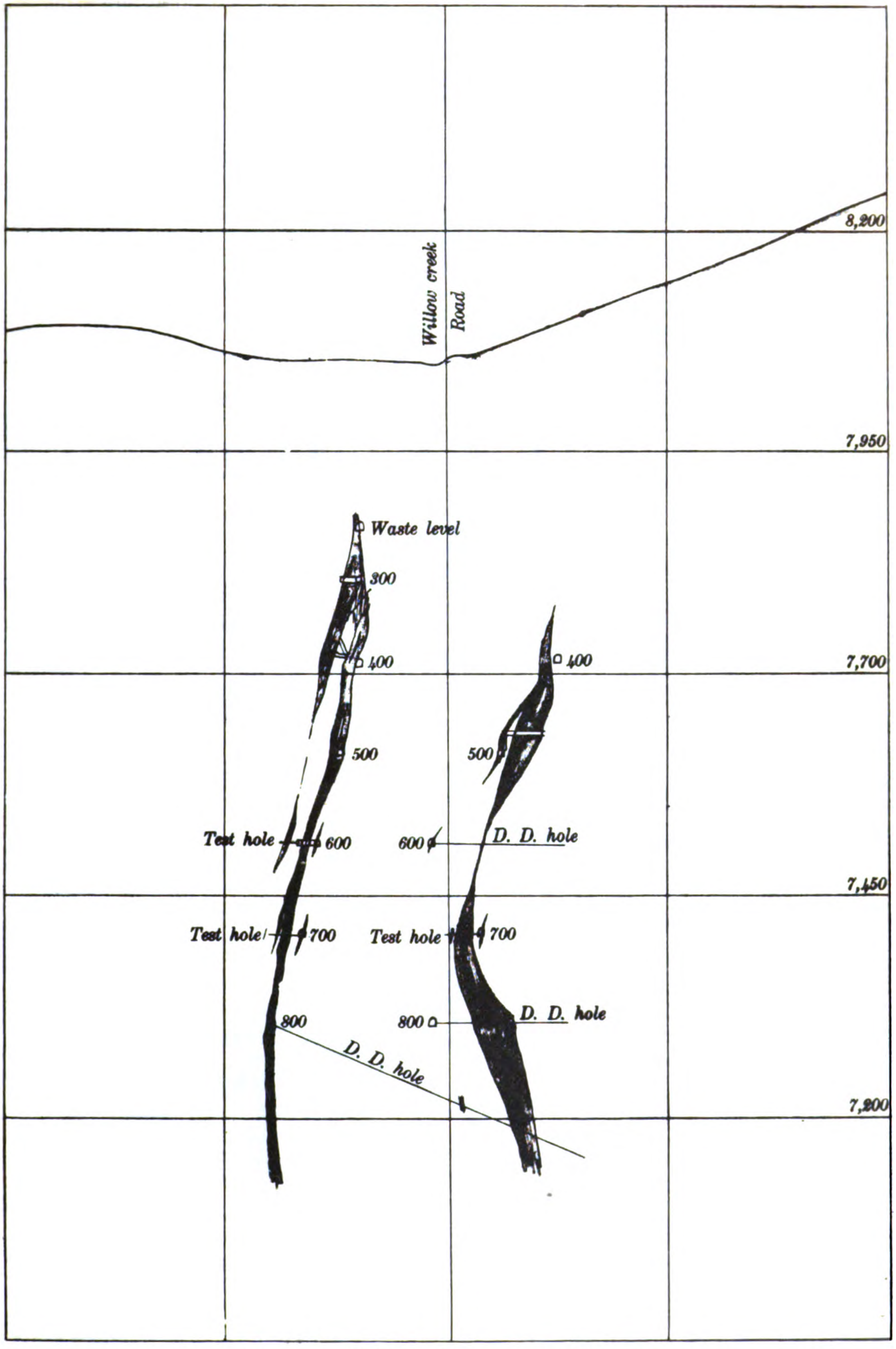


Figure 2b.– Vertical section on A–A (Figure 2a)

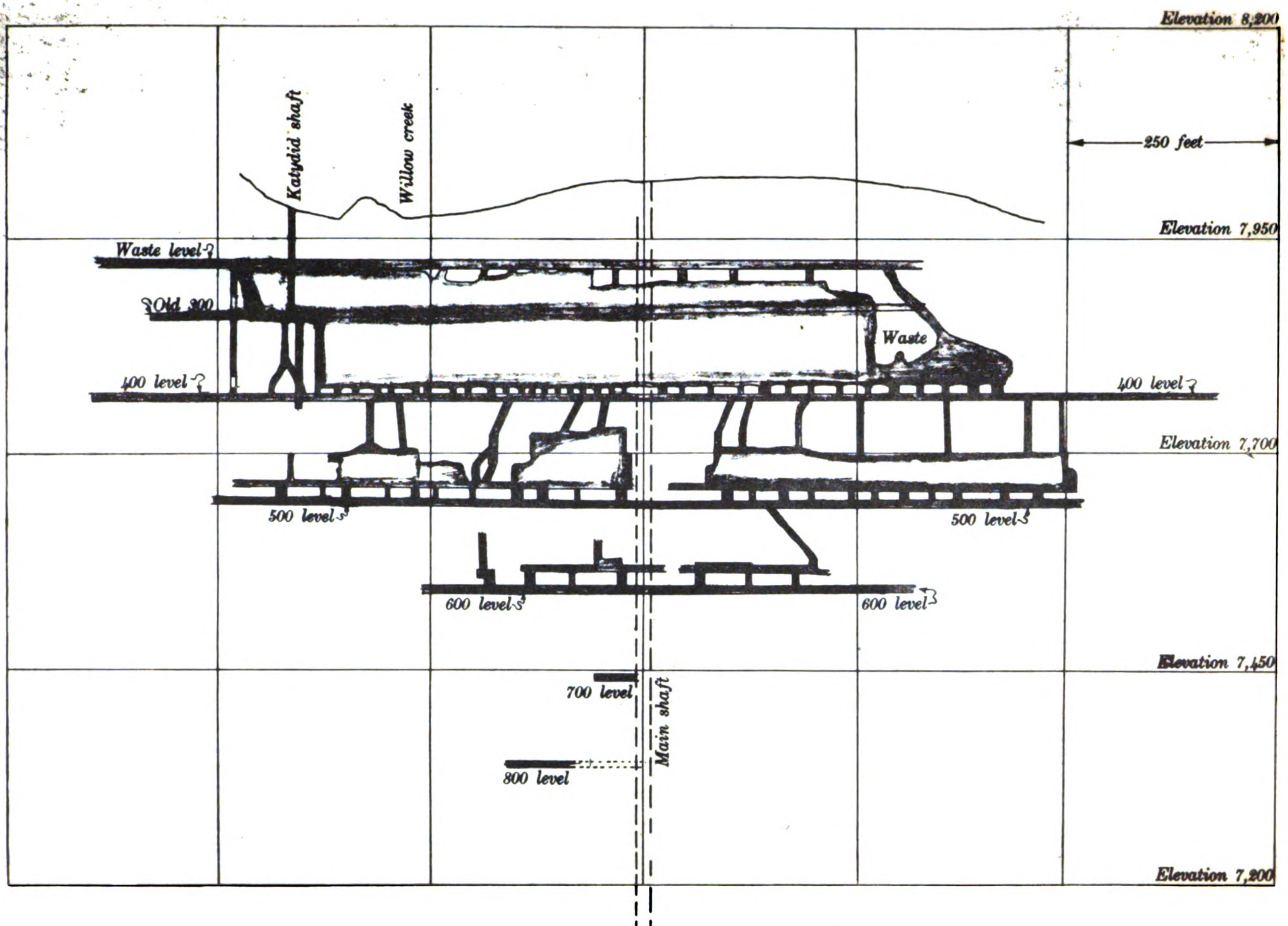


Figure 3.– Longitudinal projection along the Katydid vein

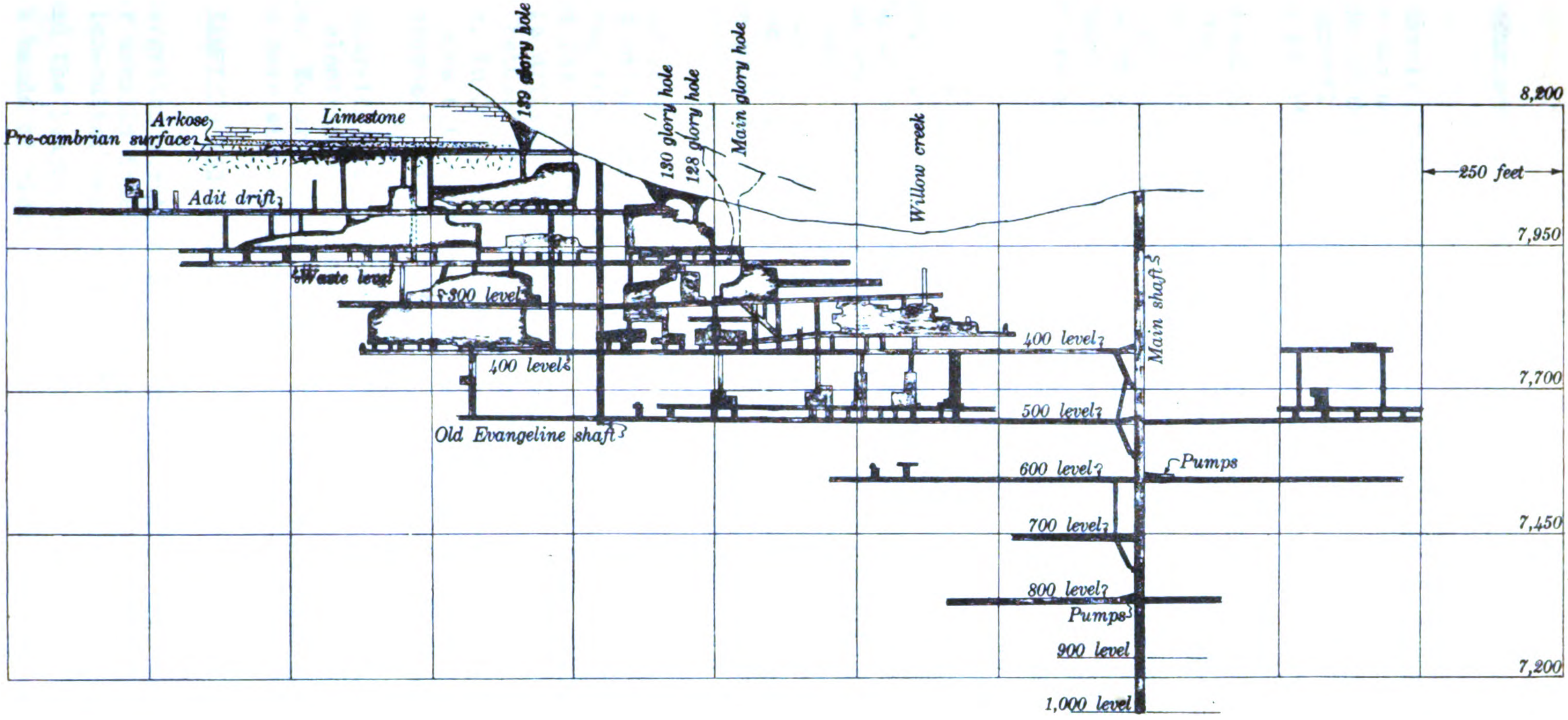


Figure 4.-Longitudinal projection along the Evangeline vein

minor parallel occurrences. Figures 3 and 4 are longitudinal sections of the main ore bodies.

The mineralization sometimes replaces all of the schist, in which case the ore body may have diorite foot and hanging walls. In other places only certain bands of the schist are replaced, and both walls may be of schist. Some movement apparently accompanied or followed deposition of the ores, but there has been no important displacement of the ore bodies.

The ore and walls are both generally very loose, and require close timbering. The ground in many places runs freely, and serious caves are likely to follow. If any initial movement can be prevented, however, support is not difficult, as the ground is neither heavy nor swelling when dry. The presence of water makes the ground very difficult to support, because of its talc content. For this reason it is important to provide drainage by developing levels below the stoping areas.

PROSPECTING AND EXPLORATION

Prospecting by drifting consists of following the principal ore shoots along their strike. If the ore body pinches out, an attempt is made to follow the most promising of possibly several sheared zones. This is a matter of great difficulty, as the ore occurrences are erratic, and no sure guides to ore are known. For this reason ore bodies when once found are sometimes explored by driving ahead the sublevel, which constitutes the sill floor of the stopes, considerably in advance of the main level drift. Following this work the main level drift can be placed to much better advantage with respect to turns, widenings, or pinching of the ore, or to parallel ore bodies.

The shear zone is crosscut or drilled at frequent intervals. The walls and downward extensions are explored by diamond drilling from underground, and extensions along the strike are explored by diamond drilling from the surface. An E-S bit is used for this work. The walls are also tested for parallel ore shoots by deep-hole drilling with Leyner machines and sectional steel. These holes are usually between 50 and 100 feet in length. The steel used is 1 1/4 inch, hollow round, in 3, 6, and 9 foot lengths. Only two-bit gages are used, as this was found more satisfactory than the several that were used originally. The cost of this prospect drilling is shown at the bottom of Table 3.

Some geophysical prospecting has been done, using the high frequency equipotential and electromagnetic methods. The results of this work were decidely unfavorable. No ore bodies were found where indicated by the surveys, whereas ore bodies have since been found in areas indicated as barren.

SAMPLING AND ESTIMATION OF TONNAGE AND GRADE

All underground openings showing mineralization are sampled by cutting channels. This is usually done with a hammer and moil, although in hard ore a compressed-air hitch-cutter may be used. Channels are cut normal to the schistosity where possible, and the length of channel included in individual samples is made to correspond with bands of high, medium, or low-grade ore or bands of waste.

The grade of ore is generally easily estimated by inspection, and sampling is unnecessary for stoping purposes. Sampling is usually done along the boundary lines of each floor of a stope previous to filling. The tonnage produced by each stope is determined by survey and checked by the car tally, which first is corrected to agree with the tons milled. No record is kept of the grade of ore produced by individual stopes. This could be done only by taking grab-samples of cars, which would probably be inaccurate with ore of this character; it is believed that all other purposes are better served by the samples taken from ore in place in the stopes.

In diamond drilling a core recovery of about 75 per cent is usually obtained. The sludge is always assayed, but experience shows that sludge assays are not reliable. Therefore when the sludge assay shows mineralization the core is split; one half is assayed and the other half is kept for a permanent record. This procedure prevents overlooking of low-grade mineralization in the drill cores. All cores are stored in core boxes in a building devoted to that purpose, where they are readily available for inspection.

Ore reserves are computed from closely spaced vertical transverse sections. Grade estimates are based on the average of all assays available in a given block. No account is taken of probable or possible ore. Therefore the ore outline is extended only a short distance into unknown ground.

METHODS OF DEVELOPMENT AND MINING

General Development

The mine is opened by two shafts and two adit levels. The main shaft, a verticle four-compartment shaft, with levels at 100-foot intervals, is midway between the Katydid and the Evangeline veins. This shaft has been sunk 50 feet below the 1000 level. All ore and waste hoisting is done through this shaft, as well as part of the handling of men and supplies. The Evangeline shaft, 1000 feet northeast of the main shaft, was sunk by the old company, and all former development work was done through it. It is a vertical two-compartment shaft, and is now used for handling men and supplies.

Horizontal development work consists of crosscuts from the shaft to the vein and of drifts along the vein, generally in the footwall. Figure 2 illustrates a typical level. Raising consists of chutes and manways from the level to the sill floor of the stopes, 20 feet above, and fill raises driven from level to level upon starting the initial stope section in a new ore body.

Usually no effort is made to obtain speed in development work, as the rate is dependent on the ease of waste or ore disposal. Above stoping sections development is carried on whenever waste is needed for stope filling. Lower level development depends on the amount of ore or waste that can be hoisted in the main shaft in addition to men, stope ore, and supplies.

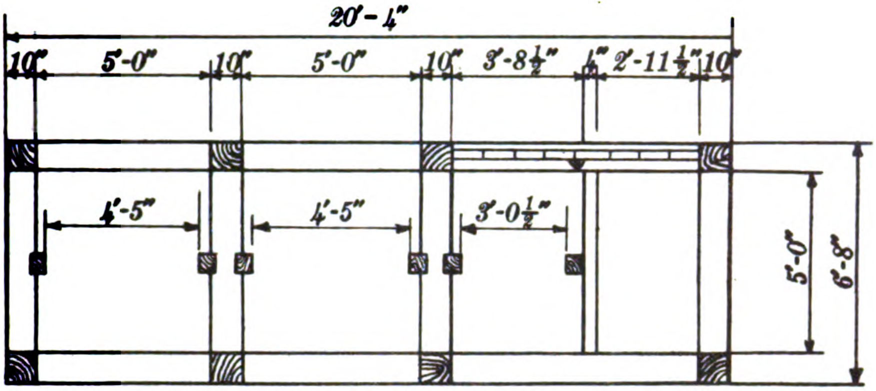
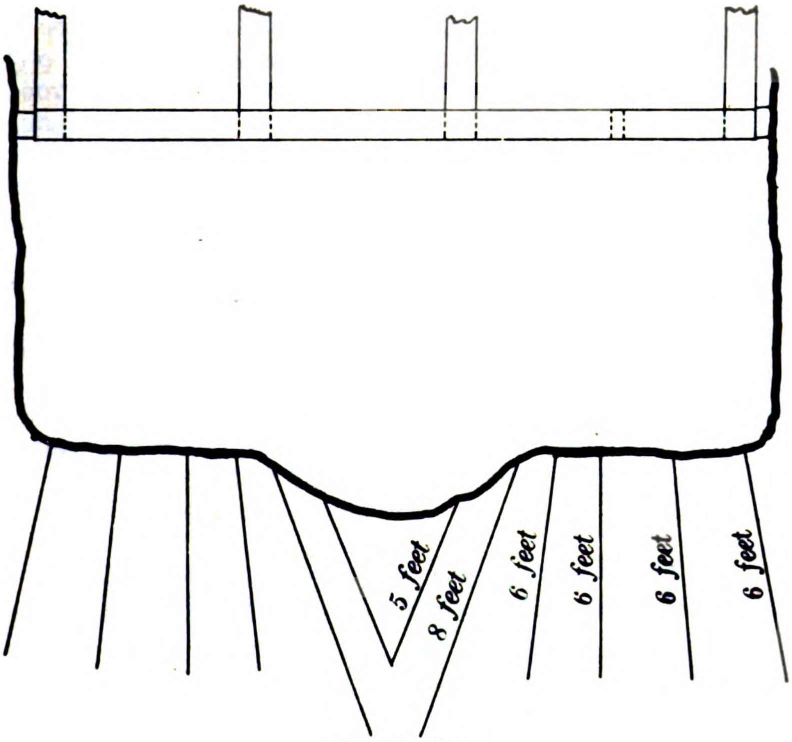
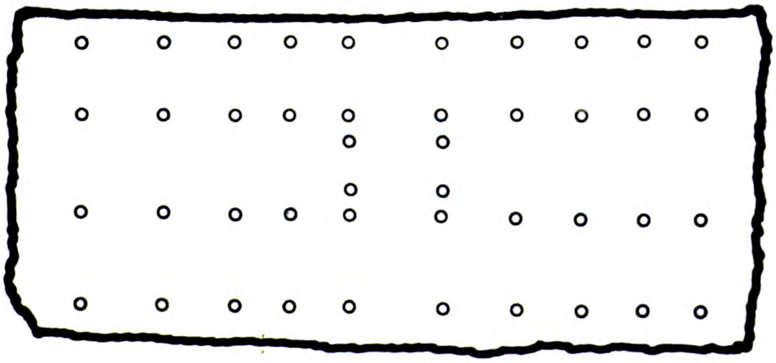


Figure 5a.—Plan of main shaft



SECTION



PLAN

Figure 5b.—Typical shaft round

Development Details

Shaft Sinking

The main shaft consists of two 5 by 5 foot skip compartments, one 4 by 5 foot sinking compartment, and one 3 by 5 foot ladder and pipe compartment (fig. 5 a). When the shaft is being sunk the two main skip compartments are bulkheaded about 20 feet below the lowest working level, and broken rock is allowed to accumulate on the bulkhead. The sinking compartment is lagged off from the skip and manway compartments from the surface to the bulkhead. A sinking bucket and safety crosshead are installed. The bucket is of the type with bail attached at the top, and has a capacity of 15 cubic feet. Two buckets are in use, so that shoveling is not interrupted by hoisting. Broken rock from sinking is hoisted to a pocket on the 200 level from which it is drawn off for stope filling. The bucket is dumped by fastening a chain, hung from a wall plate, into a ring at the bottom of the bucket, and then lowering the bucket which tips it onto a trap door. The latter is hinged to a wall plate so as to swing into the shaft at an angle of 40°.

The shaft round is of the center V-cut type (fig. 5 b). The usual round consists of 44 holes. From experience a large number of holes is known to give the best results. The rock is broken smaller, which eliminates blockholing, and less damage is done to the timber on account of the smaller amount of powder per hole. Because of the loose nature of the ground, timber must be kept close to the bottom.

An 85-pound jackhammer is used for sinking. The heavy machine has been adopted because at times ribs of hard rock are encountered which heretofore have required the use of mounted Leyner machines. Drill steel is 1 1/8-inch hollow round, lugged for Leyner chuck.

Rounds are blasted with two to four boxes of 40 per cent strength special low-freezing gelatin dynamite. Firing is done with 10 delays of No. 8 electric delay blasting caps, using the 440-volt circuit. Connections to the bottom are made through several hundred feet of blasting wire carried on a reel. For safety the shaft boss keeps the key and the fuse plugs of the switch in his pocket. Connections at the bottom are made to two lead wires suspended on stakes clear of the ground. All black wires are secured to one lead, and the red wires to the other. The leads are not connected to the blasting line until the men are ready to leave the shaft.

During shoveling, drilling, and timbering light is furnished by several electric bulbs in a large reflector. Before loading starts the light line is disconnected and raised to avoid possible stray currents from this source.

Usually a 6-foot round requires 8 hours for extending pipe lines, drilling, loading, raising the pump, and blasting, 6 hours for timbering, including guides and ladders, and 24 hours for shoveling, including lowering the pump. Shoveling is difficult because the broken rock tends to cement itself, and picking is required for each shovelful.

Coeur d'Alene lagging, that is, lagging held by cleats on the top and bottom of wall and end plates, is used because it strengthens the sets when blasting is done close to timber. It has of course the advantage of being easily replaced. About 50 gallons of water per minute is pumped from the shaft in addition to the amount hoisted during mucking. Concrete water rings are placed about every 100 feet in the shaft. These serve as bearers, as well as to catch water coming down the shaft walls and timber. The water caught on the ring is raised to the nearest level above, usually not over 20 or 25 feet, by means of a water eductor. This is a device similar to an injector, using water under pressure as a motive force.

Air operated displacement pumps are used in the bottom of the shaft. The air exhaust is piped into the water column, to avoid freezing of the exhaust valves and also to eliminate noise. Two pumps are used, and interchanged after each round. The one not in service is kept on the first level above the bottom, where it is thoroughly inspected as to packing and repairs before being lowered agai
In addition an electric vertical centrifugal pump of larger capacity is kept in reserve in case a larger water flow should be encountered.

Drifts and Crosscuts

Crosscuts from the shaft to the veins are driven 7 by 8 feet in section. Haulage drifts are timbered 7 feet high, 5 feet, 4 inches wide at the top, and 5 feet, 10 inches at the bottom in the clear. Figure 6 a shows the timber drift set used. The haulage drifts are usually driven in the ore following the footwall. A toe-cut round drilled from a vertical column is used in drift work. Twelve to 16 holes are drilled for a round, but in the loose ground which generally prevails the back holes are not blasted. If necessary, they can be blasted later to make room for timber. This avoids many serious runs or caves.

All shoveling is done by hand. No regular daily cycle of operations has been found possible because of the very irregular nature of the ground. Consequently the crews on both shifts drill, shovel, or timber, as may be necessary.

Raises

From the main haulage drifts cribbed manways and chutes are driven to the stope sills about 20 feet above the level, at intervals of 25 to 60 feet, depending on the width of the vein. The raises are cribbed with round timber about 6 inches in diameter (fig. 6 b), forming two compartments each 4 by 4 feet in the clear. Short pony sets are placed over the drift sets where a raise is to be started. The pony set caps are long, extending to solid ground at one side of the drift. These caps form the sills on which the raise cribbing is placed.

Fill raises from the stope sills to the level above are generally stulled, about 4 by 6 feet in section. Three-inch plank is nailed to one side of the center stulls to form a manway. In some places these raises must be cribbed, in which case they are larger, with two 4 by 4 foot compartments.

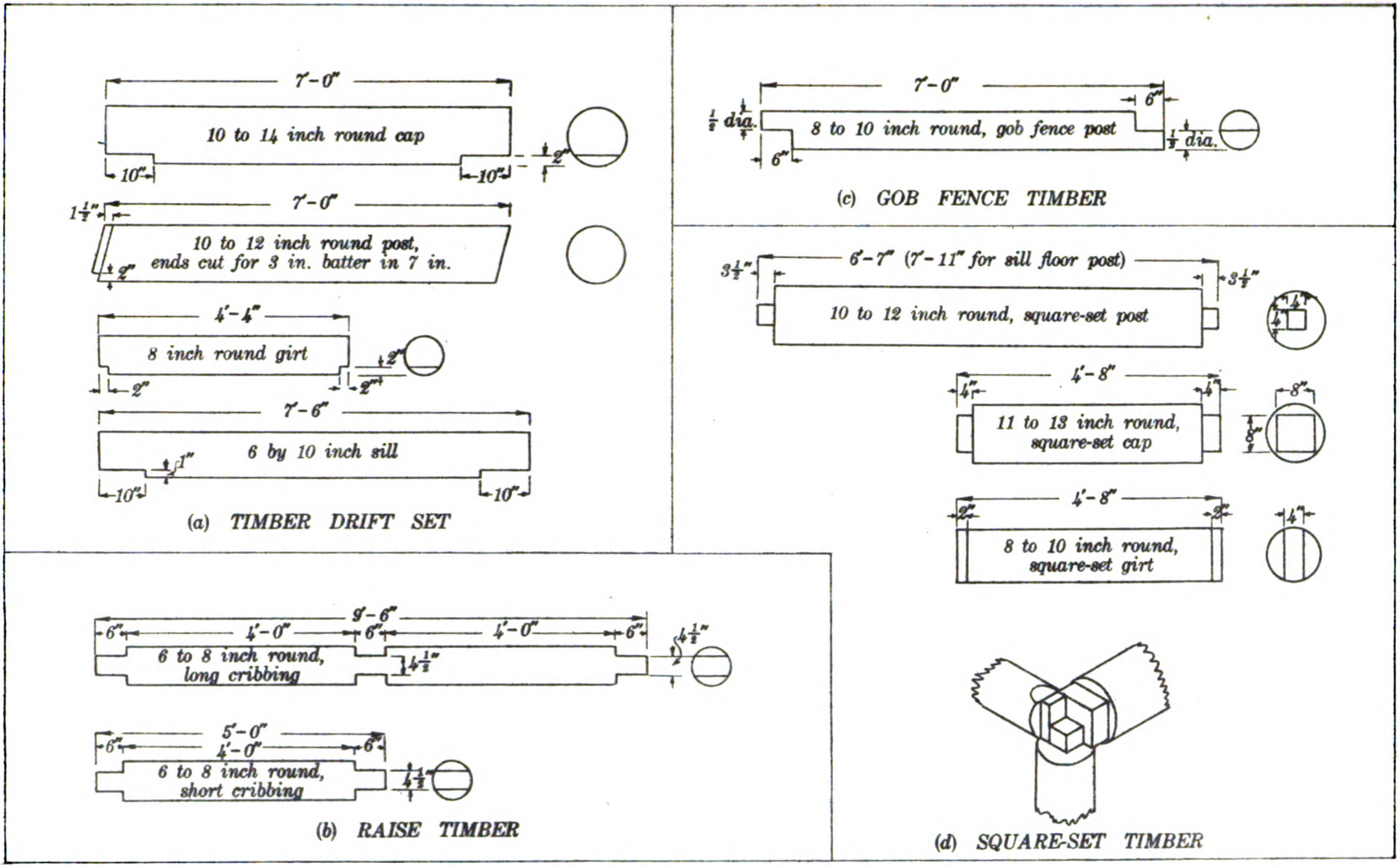

7'-0"
10 to 14 inch round cap
10"
10"
2"
7'-0"
1½"
10 to 12 inch round post, ends cut for 3 in. batter in 7 in.
2"
4'-4"
8 inch round girt
2"
2"
2"
7'-6"
6 by 10 inch sill
1"
10"
10"
(a) TIMBER DRIFT SET
9'-6"
6"
4'-0"
6"
4'-0"
6"
6 to 8 inch round, long cribbing
4½"
4½"
5'-0"
4'-0"
6"
6"
6 to 8 inch round, short cribbing
4½"
(b) RAISE TIMBER
7'-0"
6"
⅛ dia.
8 to 10 inch round, gob fence post
6"
⅛ dia.
(c) GOB FENCE TIMBER
6'-7" (7'-11" for sill floor post)
3½"
3½"
10 to 12 inch round, square-set post
4"
4"
4'-8"
4"
4"
11 to 13 inch round, square-set cap
8"
8"
4'-8"
2"
2"
8 to 10 inch round, square-set girt
4"
(d) SQUARE-SET TIMBER

Figure 6.-Mine timber

Drilling

Three types of drills are employed in the mine. Leyner-type drifters are used in drift work and sometimes in stope sills, mounted on 3 1/2-inch bars. Heavy jackhammers are used for shaft sinking and lighter ones in the stopes for breaking ground and blockholing. Hand-rotated stopes are used for raise work and in stopes.

A repair shop for drills and hose is near the collar of the main shaft. A "drill doctor" visits all working places daily to attend to minor repairs and to keep track of the machines.

Three kinds of hollow drill steel are used: 1 1/8-inch round for Leyners and shaft-sinkers, 1-inch quarter octagonal for stopers, and 7/8-inch hexagonal for light jackhammers. A single 14° taper cross bit is used for all purposes. Starter bits are 2 1/4 inches in diameter, and gage changes 1/8 inch. Length changes are 18 inches for Leyner and jackhammer steel and 12 inches for stoper steel.

During 1929 an average of 360 pieces of drill steel were sharpened daily. This was done by one drill sharpener who also did the tempering, and two helpers.

A 3,000 cubic foot per minute compressor near the shaft collar supplies air at 100 pounds pressure, which is sufficient to maintain a pressure of 85 to 95 pounds at the drills.

Drilling water is piped into all working places. This water is from the camp domestic water line.

Blasting

Two kinds of explosives are used underground, 40 per cent strength special low-freezing gelatin and 25 per cent strength straight gelatin dynamite. The main magazine is on the surface, and only enough powder for daily needs is kept underground. The nipper on each level has charge of the magazine and distributes the powder to the working places.

Number 8 electric delay detonators are used in all wet places, in high raises, and in long straight headings. In relatively dry places No. 6 tetryl caps are used, with triple-taped waterproof fuse. About half of the tetryl caps are treated with cap-sealing compound.

No stemming is used, and each miner does his own blasting.

Stoping

The conditions governing the selection of stoping methods at the Pecos mine are as follows:

1. The ore shoots are irregular and not continuous, pinching out and giving way to others. Sometimes two to four parallel veins are separated by only

10 to 20 feet of barren, loose schist or blocky diroite. Occasionally the parallel veins can be mined simultaneously, leaving the waste in place, but usually the latter must be mined and gobbed back in the stope when the ore is mined.

2. As the veins contain many low-grade or barren schist bands and diorite intrusions, the mining method adopted must provide for sorting waste in order to avoid serious dilution. It is estimated that at least 20 per cent of the stoping ground broken is sorted and rejected in the stope as waste.

3. The method must be such that runs from the walls can be checked quickly.

4. It must be possible to prospect the walls at close intervals for parallel ore shoots.

5. Faulting accompanying or following the ore deposition has left gouge 1 to several feet thick which must be guarded carefully to prevent runs.

6. Drainage must be provided before stoping can begin.

7. The surface must be kept intact. The Pecos River, with a normal flow of 200 second-feet, parallels the outcrop of the veins at a distance of only 400 feet, and the dip of the veins is toward the river. Willow Creek, with a flow of 20 second-feet, crosses the shear zone at a right angle at its point of greatest width.

8. The complex ore must all be milled, and only one class of ore has to be handled. The ore apparently is not harmed for milling purposes by standing in the stopes after breaking.

The cut-and-fill and the square-set-and-fill methods are now used about equally, as shown in Table 3, at the end of this paper. The choice between them depends on the character of the ore and walls where stoping is started. Strong walls and ore favor the cut-and-fill method. Many times, however, it is possible or necessary to change from one method to the other after stoping has been carried up several floors.

Pillar mining by underhand stoping similar to that used at the Magma mine, Superior, Ariz.,[4] has been used successfully, with an average cost per ton slightly below that of square-setting. However, experience at this mine has shown that very close supervision is required when the pillar mining method is used. Having such a large unfilled area, the stopes are caved easily in the event of a slight misjudgment on the part of miners or bosses, and under such conditions a caved stops is much more expensive to reopen than a filled stope.

Shrinkage stoping has been possible in only one stope, near the top of the ore body. The ore and walls there were stronger than usual, and due to its position caving would have no serious effect on future operations.

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4 - Snow, Fred W., Mining Methods and Costs at the Magma Mine, Superior, Arizona; Information Circular 6168, Bureau of Mines, 1929, 32 pp.

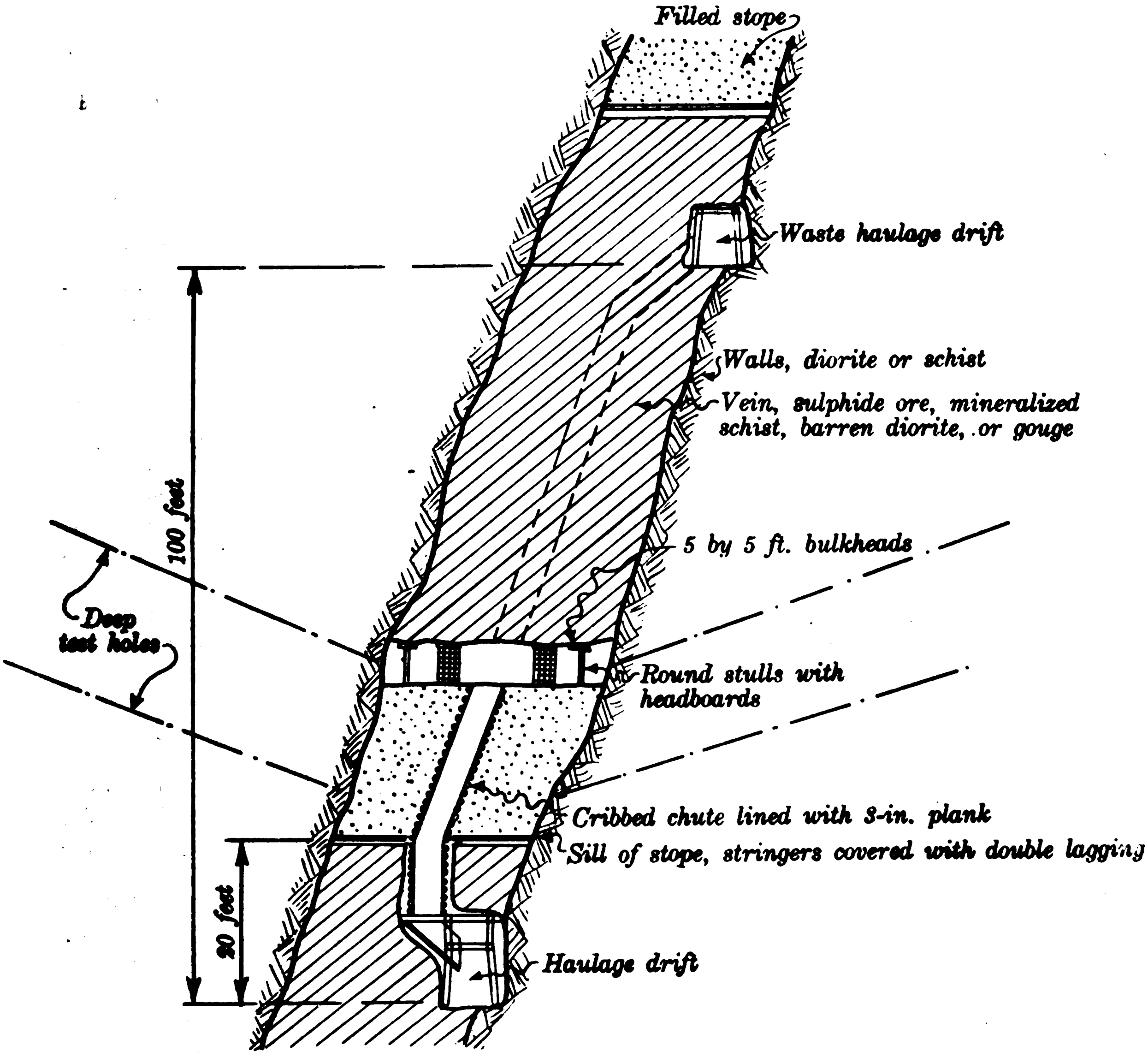

Filled stope
Waste haulage drift
Walls, diorite or schist
Vein, sulphide ore, mineralized schist, barren diorite, or gouge
5 by 5 ft. bulkheads
Round stulls with headboards
Deep test holes
100 feet
Cribbed chute lined with 3-in. plank
Sill of stope, stringers covered with double lagging
20 feet
Haulage drift

Figure 7. — Section across vein through cut-and-fill stope

The inclined cut-and-fill method has been tried and rejected for several reasons: The tendency of the ore and walls to cave and run, with consequent flooding of the stope and danger to workmen below; the difficulty of erecting stulls, cribs, or bulkheads on an inclined floor; the lack of storage space for waste. These disadvantages offset the saving effected by gravity-handling of ore and waste on the inclines.

Horizontal cut-and-fill

From the main haulage drifts 2-compartment raises are driven 12 feet above the back of the drift, and the stope is silled out the full width of the ore (fig. 7). Ground conditions as found at this point determine the choice between cut-and-fill and square-set methods. The horizontal cut-and-fill is used when possible, even though in most cases the ore and walls are so loose that many stulls and bulkheads are required. The timber consumption in fact is often the same as for square-setting. However, timbering in a square-set stope requires more skilled labor, and a substantial saving in labor cost is thus effected by cut-and-fill stoping.

The ore bodies are mined out in sections, the length of a section depending on the width of the vein and on ground conditions. During the first two years of operation, in wide veins, stope sections were from 15 to 60 feet long, with 15-foot pillars between. More recent practice, in new wide-vein areas, is to cut initial stopes 30 feet long, spaced 100 to 150 feet apart, with extraction chutes in both ends of the sections. After these initial stopes have reached or nearly reached the level above, new sections, usually about 15 feet long, are started on one or both sides of them, using the former extraction chutes for fill raises. Pillars occur only where two stope blocks come together.

The stopes are mined upward by taking successive horizontal, 7-foot cuts from hanging to foot wall, following each cut by filling.

Stope end lines are timbered off from the solid by means of vertical gob posts (fig. 6 c), spaced 5 feet apart and laced with lagging. The posts are tied back into the fill by means of old cable.

In narrow veins stopes are silled several hundred feet in length, with chutes spaced 30 feet apart. Every other chute is provided with a manway. Fill raises are driven in ore 30 or 60 feet apart depending on the width and grade of the ore.

Stopers are used for drilling in stopes where possible, but in many places the ground is so loose that it is more satisfactory to drill flat holes with jackhammers. Also because of the nature of the ground it is necessary to provide prompt support for the back, and breaking ground resolves itself into making room for timbers, rather than to producing tonnage. Little heavy blasting is done, and blasting is not concentrated at the end of the shift.

Before blasting, a shoveling floor of 2-inch plank is laid on the fill. This plank is used again on the next floor, or if broken, for lacing the outside of the cribbed raises. All shoveling is done by hand because of the necessity for sorting.

The cribbed extraction chutes are lined inside with 3-inch plank. The lining is repaired after each cut in the stope, and often requires repair during a cut, to prevent runs of waste fill. The tops of the chutes are covered with grizzlies of round timber spaced about 10 inches apart.

Back and floor pillars are left until all stoping is completed on the level below. They are then removed by square-setting in short sections, usually placing two sets along the vein and extending them from wall to wall. The general rule in this work is to retreat toward natural outlets.

As a rule the drifts are kept open permanently, but they are not necessarily of standard size. Sometimes drifts which are not needed are filled tightly when the stopes are filled.

Square-Set-and-Fill Mining

The same general scheme of operations applies to square-setting, except that manways, extraction raises, and gob lines are provided by lacing off square-cuts. Square-sets are of round timber, and are 5 feet square and 6 feet, 8 inches high, excepting sill floor sets, which are usually 8 feet high (fig. 6 d). The caps are stronger than the ties, and are therefore placed normal to the strike, which is the direction of maximum pressure.

Waste Filling

Waste for filling stopes is obtained from development work on the levels, prospect workings driven into the walls from the stopes themselves, from sorting waste, or from a glory hole on the surface.

A raise from the upper level known as the waste level is connected to a glory hole at the surface. A bulldozing chamber with a 12-inch grizzly is located in the raise about 40 feet above the level. During the last two years, however, prospect and development workings have furnished sufficient waste for all needs, and the glory hole has not been used.

Transportation

Underground Haulage

All tracks are of 16-pound rail and 24-inch gage.

Ore cars are of the rocker-dump type, having a capacity of 40 cubic feet (fig. 8). In development work in waste a side-dump car of 18-cubic-foot capacity is used (fig. 9). This car is small enough to go on the cage. The reason for the use of a side-dump car is connected with future waste haulage. It is foreseen that by the time some of the present levels are in extensive use for waste haulage, many of the drifts will have squeezed enough to seriously decrease the clearance. Under such conditions the side-dump car will be more convenient than the end-dump type, which must be uncoupled and turned in order to dump.

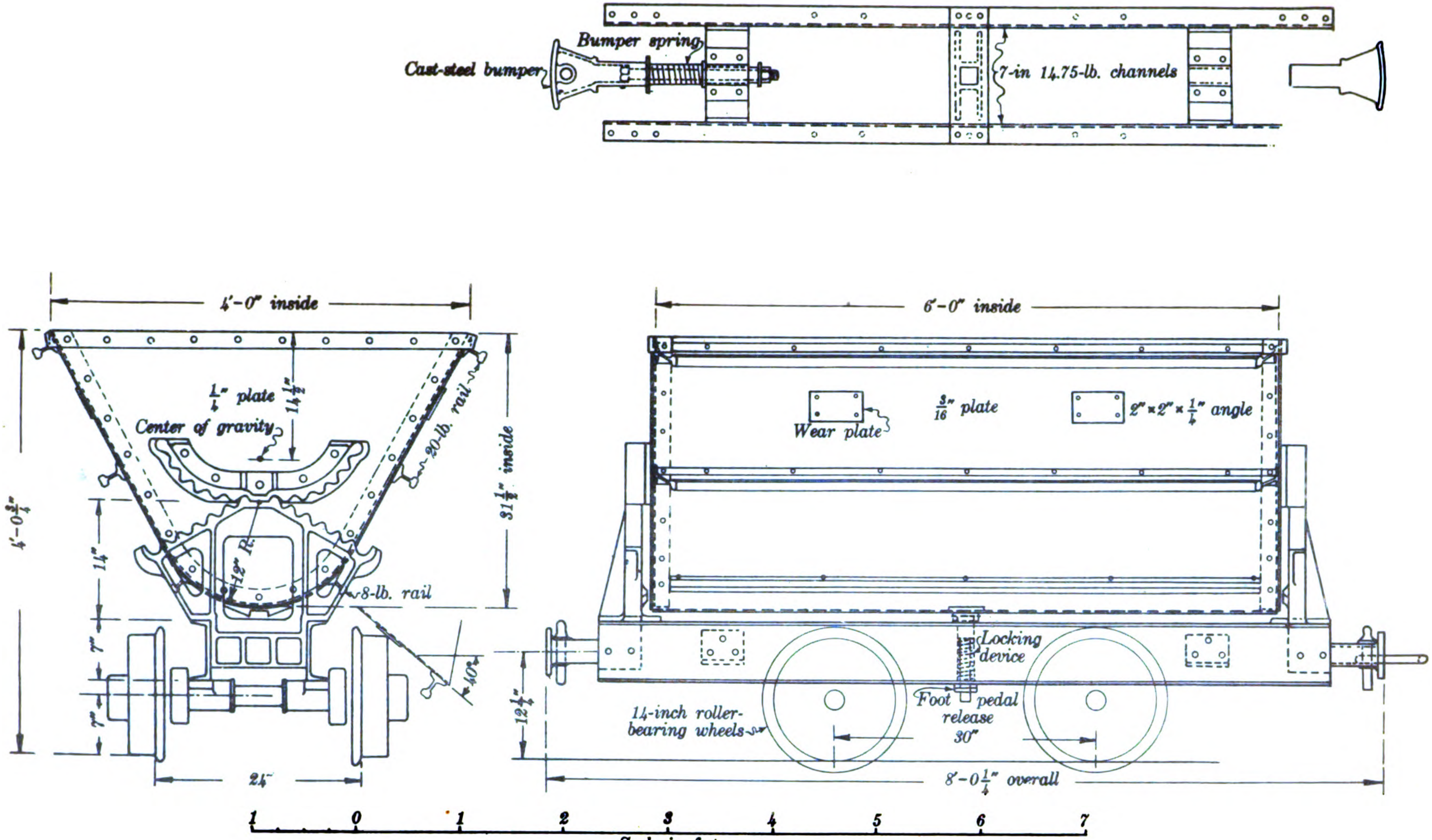


Figure 8.—40-cubic feet rocker-dump car

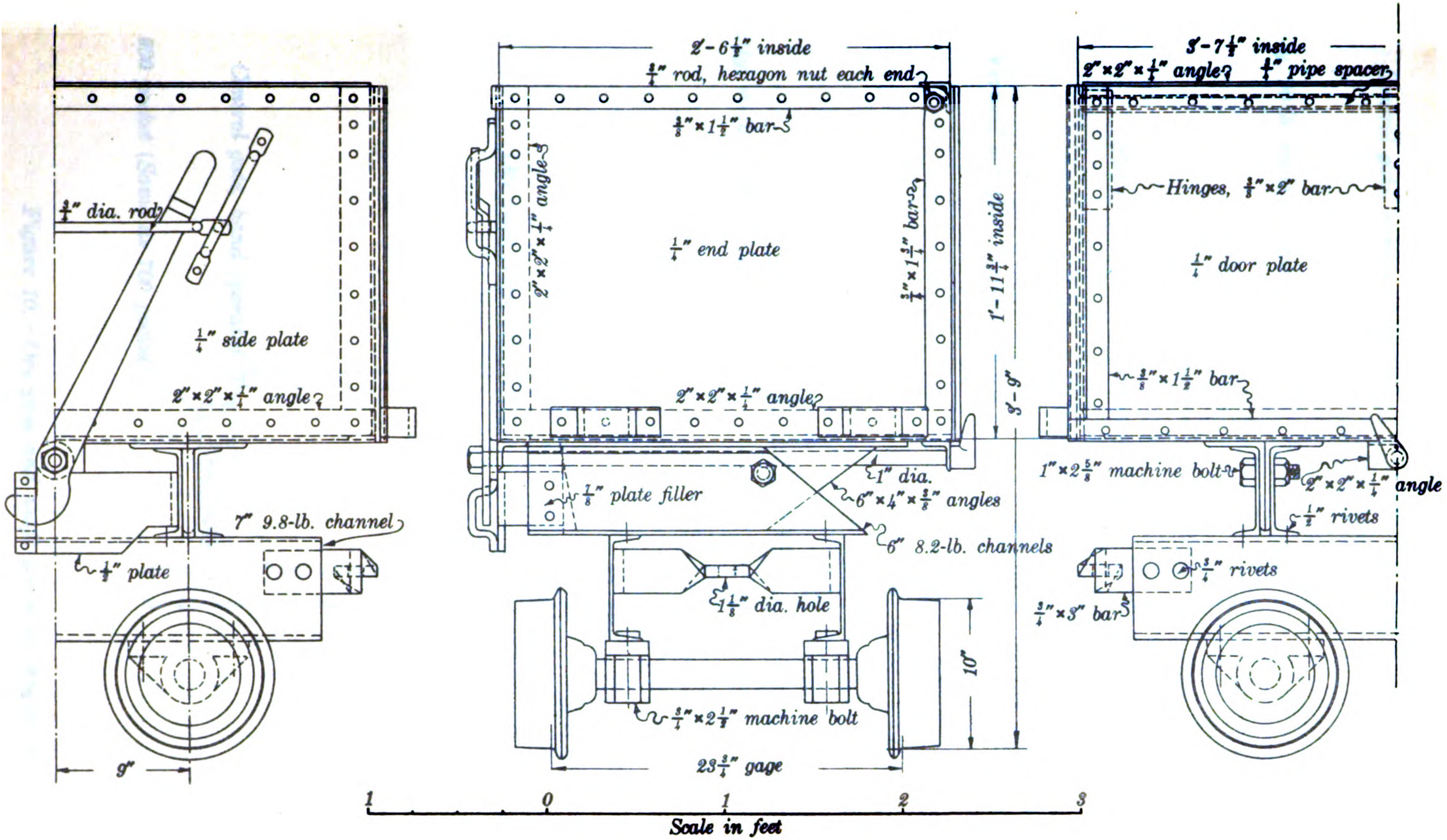


Figure 9.–18 cubic feet, side-dump car

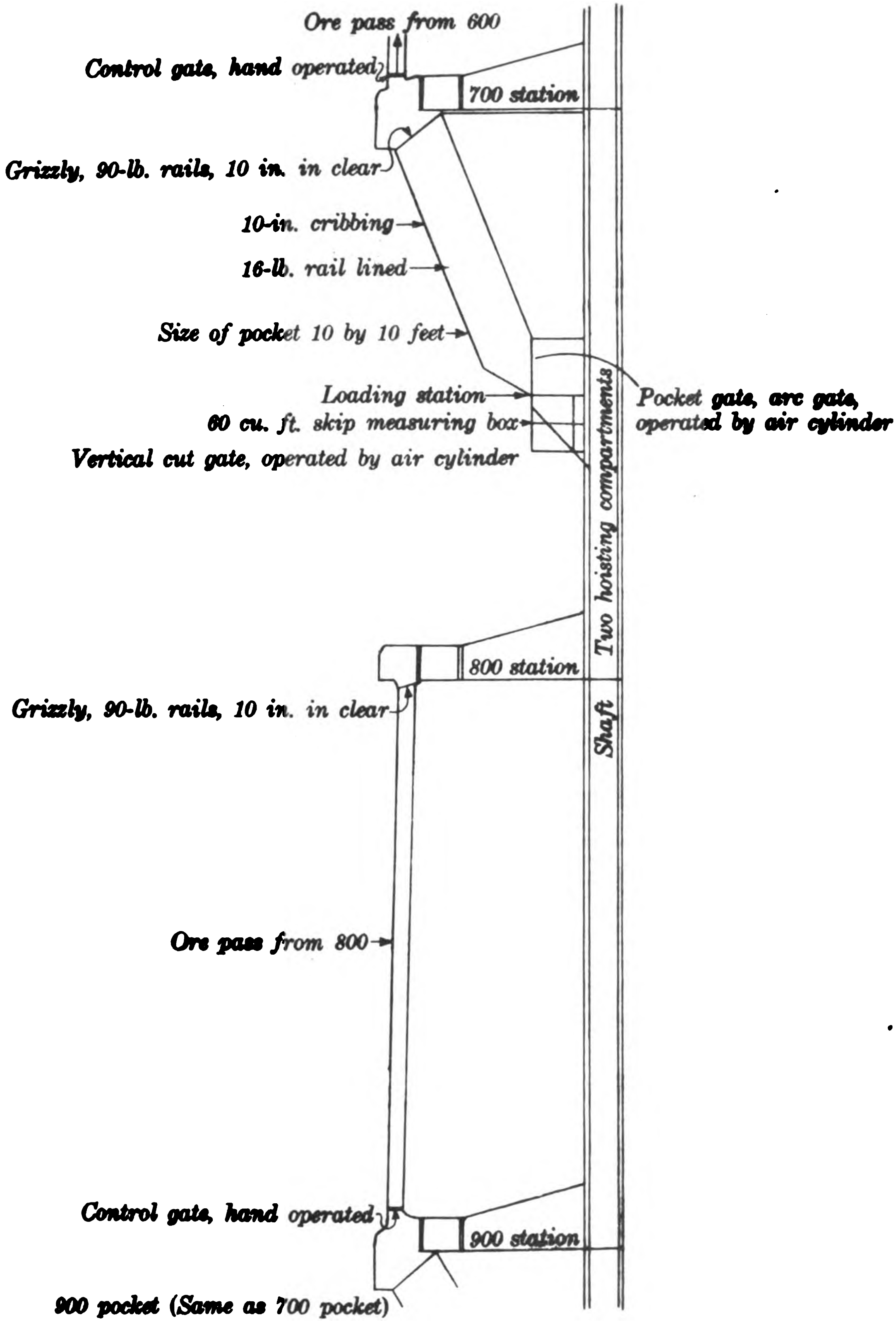

Ore pass from 600
Control gate, hand operated
700 station
Grizzly, 90-lb. rails, 10 in. in clear
10-in. cribbing
16-lb. rail lined
Size of pocket 10 by 10 feet
Loading station
60 cu. ft. skip measuring box
Vertical cut gate, operated by air cylinder
Pocket gate, arc gate, operated by air cylinder
Two hoisting compartments
800 station
Shaft
Grizzly, 90-lb. rails, 10 in. in clear
Ore pass from 800
Control gate, hand operated
900 station
900 pocket (Same as 700 pocket)

Figure 10.—Ore pass and shaft pocket arrangement

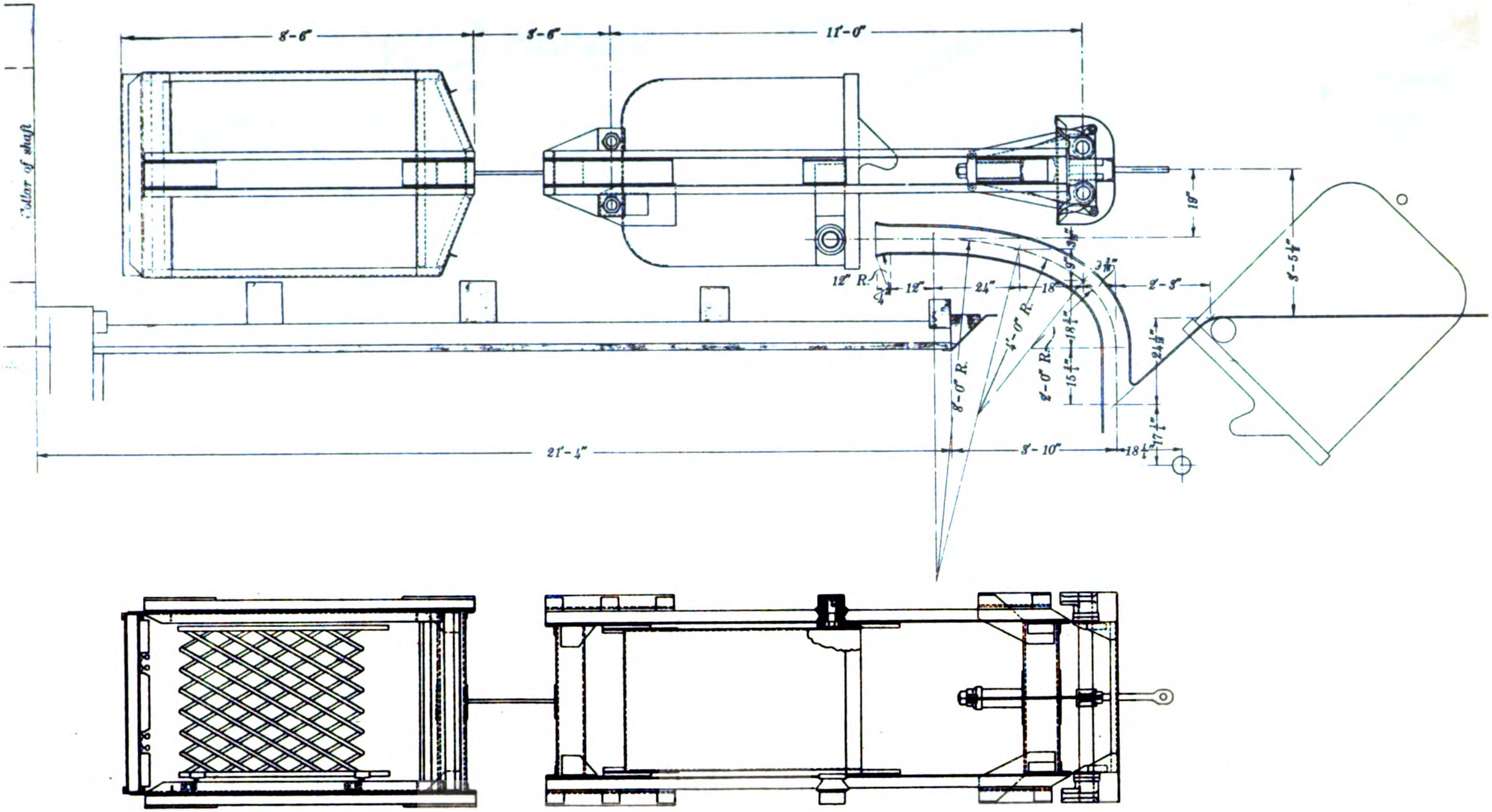


Figure 11.-Assembly of cage, skip, and skip dumping arrangement

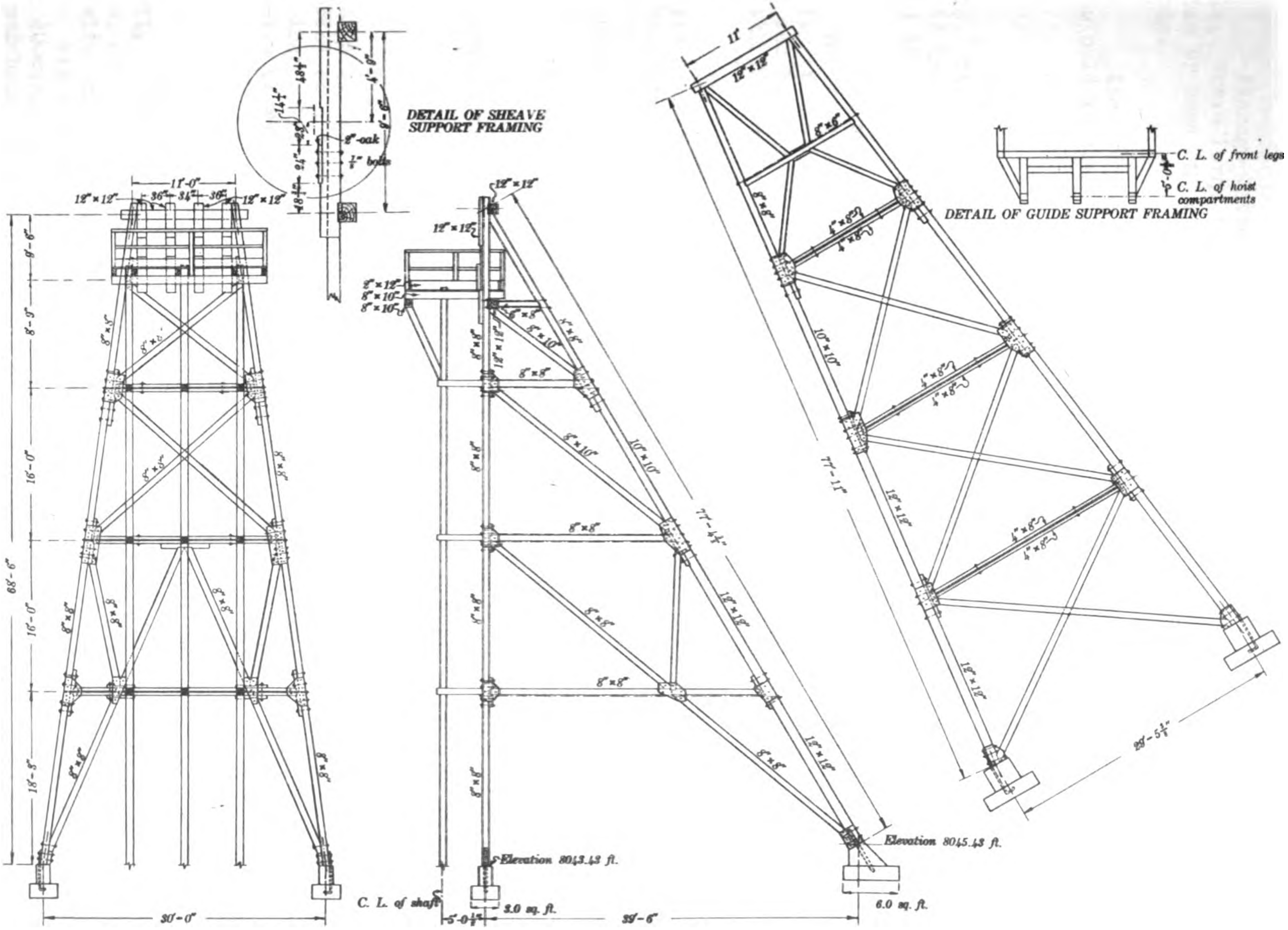


Figure 12.—Shaft head frame of timber construction

Storage-battery locomotives are used on the 300, 400, 500, and 600 levels. Where large tonnages are handled, two batteries are provided. Tramming is done by hand on new levels until the distance from the shaft warrants motor haulage.

Stope chutes are hand-operated, of timber construction, with chute boards held in cleats.

Hoisting

Ore pockets have been cut at the main shaft on the 500, 700, and 900 levels. Other levels are served by by-pass raises, as shown in Figure 10. The pockets have a capacity of 300 tons each. Steel grizzlies of 90-pound rail spaced 10 inches apart are placed over the pockets.

The skip compartments of the main shaft are equipped with combination skips and cages, having the cages hung permanently underneath (fig. 11). The capacity of the skips is 3.5 tons. They are operated in balance. The rope speed for both ore and men hoisting is 720 feet per minute. Hoisting capacity from the 700 pocket is about 100 tons per hour.

The main shaft sinking compartment is equipped with a single-drum electric hoist with a rope pull of 12,000 pounds. When sinking is not under way this hoist operates a small skip which handles waste from development work. This waste can be dumped into the waste pocket above the 200 level or into the surface waste bin, from which it is loaded into cars and trammed to the dump.

Figure 12 shows the timber headframe of the main shaft.

The Evangeline shaft is provided with a single-drum electric hoist of 7,000 pounds rope pull, operating a single-deck cage.

CRUSHING PLANT

The ore bin at the collar of the main shaft has a capacity of 100 tons. From this bin the ore goes to an 18 by 30 inch jaw crusher. After crushing, some waste is sorted on a conveyor belt and then the ore goes to a 10-inch gyratory and 20 by 54 inch rolls. The resulting 1-inch product is taken by a conveyor belt to a 1,500-ton storage bin at the tramway loading station.

AERIAL TRAMWAY

The aerial tramway structures consist of terminal stations at the mine and mill, a control station 4 miles below the mine, 15 tension stations, and 105 towers, all of steel construction. The track cable on the loaded side is 1 3/8-inch size and on the light side is 1 inch. The traction rope is 3/4 inch in diameter. The tramway is operated by a 100-horsepower motor at 500 feet per minute. Using 10-cubic-foot buckets spaced 350 feet apart the line has a capacity of 50 tons per hour. The 12 miles of line includes several spans over 4,000 feet long.

MINE DRAINAGE

At present an average of 1,200 gallons of water per minute is being pumped from the mine. A pumping station on the 500 level contains one 500 gallon per minute centrifugal pump and one 460 gallon per minute vertical plunger displacement pump. These bring water to the surface through an 8-inch column. On the 800 level are three 500 gallon per minute centrifugal pumps pumping to the surface through another 8-inch column. One 500 gallon and one 1,000 gallon per minute centrifugal pump are now being installed on the 1,000 level. All centrifugal pumps are direct-connected to 440-volt electric motors, excepting those on the 1,000, which will be driven by 2,300-volt motors. The plunger pump on the 500 level gave much trouble by wearing out pinion gears, with attendant bearing troubles, until a patented form of rope drive was substituted for the gears. The rope drive is giving perfect satisfaction. A 250-gallon electric-driven piston pump is installed on the 400 level of the Evangeline shaft to handle water that can not be drained to the main shaft. None of the mine water is acid.

MINE TIMBER SUPPLY

The Pecos mine is favorably situated near forests of native white pine, fir, and spruce large enough to supply mine requirements for many years.

Until recently the round peeled poles, varying from about 5 inches to 14 inches in top diameter, were hauled in by teams to the timber yard at the collar of the Evangeline shaft. The equipment there consists of a swing cut-off saw, a rip saw, and a framing machine. The small poles, averaging about 6 inches in diameter, are used for round raise cribbing, and any oversupply of such sizes is split into half-round lagging. The larger poles, from 8 to 14 inches in diameter, are framed into square-set or drift timber. Additional lagging is obtained by slabbing large poles into 2-inch plank with round edges. Thus there is no waste except the sawdust.

As payment to the Government for stumpage, and the contracts for cutting, peeling, skidding and hauling are based on linear feet for various sizes, it is difficult to make comparisons on a board-foot basis. However, it is estimated that sized timber would have to be delivered to the mine at $16 per thousand board feet to compete with the round timber purchased from the Government.

Recently a saw mill has been built in the woods, and areas closer to the mine are now being recut for large saw timber. A sufficient number of teams for hauling can no longer be obtained, and large tractors and tractor trailers have been purchased. The saw mill, with all mechanical handling equipment will deliver timber to the mine yard at less than $20 per thousand board feet.

The cost of timber is an important item at this mine, as about two-thirds of the stoping is at present done by square-setting; the haul from the nearest railroad point would be 18 miles.

PER CENT EXTRACTION

As a rule ore estimates are only approximate because of the irregularity of the ore shoots. However, nearly 100 per cent extraction is made of the known ore. Stopes are filled tightly and any caved stopes can always be recovered. The walls are explored thoroughly for parallel ore shoots. Low-grade areas generally consist of parallel bands of high-grade ore and barren waste, which can be sorted. Only in exceptional cases are blocks of low-grade ore left behind to be mined during periods of high metal prices.

WAGES, CONTRACT, AND BONUS SYSTEMS

The wage scale at the Pecos mine, which has not been changed since operations started, is as follows:

	Per 8-hour shift
Timbermen	$ 5.00
Miners	4.50
Shovelers	3.50
Trammers	3.50
Motormen	4.50
Brakemen	4.00
Cagers	4.00
Nippers	3.50
Shaftmen (wet)	8.00

The mine is a considerable distance from other metal mining districts, and about 75 per cent of the labor employed has been developed from Spanish-Americans who before the opening of the mine lived on small farms in the Pecos Valley or in small near-by towns. Records show that 65 per cent of the miners and timbermen have been trained from raw recruits who had never before worked in a mine.

About 75 per cent of the development work is done by contract. Because of loose ground conditions very few stopes are contracted. Rates per foot depend on the size of the opening and on general conditions. No cycle of operations can be planned for development work because of the irregular character of the ground, and consequently drilling, shoveling, and timbering are all covered by a single contract. The contracts are for labor only; the company furnishes explosives and all other supplies and equipment. Men on contract work are guaranteed at least regular wages. The company deducts such wages from the contract earnings and pays the men their regular wages. The balance of the earnings is paid to the contractors, who in turn pay their crews. The amount of such payments is not known to the company. However, contractors are usually reliable men who have been in the employ of the company for a considerable time, and rarely is any dissatisfaction reported regarding the method of payment.

The average contract price for untimbered drifts is $5 per foot; for timbered drifts, $8.50 per foot; for raises, $4 per foot for the first 50 feet, an $5 per foot for the remainder.

Contracts are given out by the mine foreman subject to the approval of the mine superintendent.

All bosses are paid a bonus based on tons per man-shift.

VENTILATION

Natural ventilation is excellent because of the many openings to the surface, and all workings are very cool. No mechanical ventilation is used in the mine, with the exception of a few small fans with canvas tubing employed for opening new levels, before connection is made to levels above. Gases from blasting are expelled from dead ends by means of compressed air.

In the winter months, which are extremely cold, air doors are installed in many parts of the mine to check air currents and avoid frozen air and water lines.

FIRE HAZARDS AND SAFETY METHODS

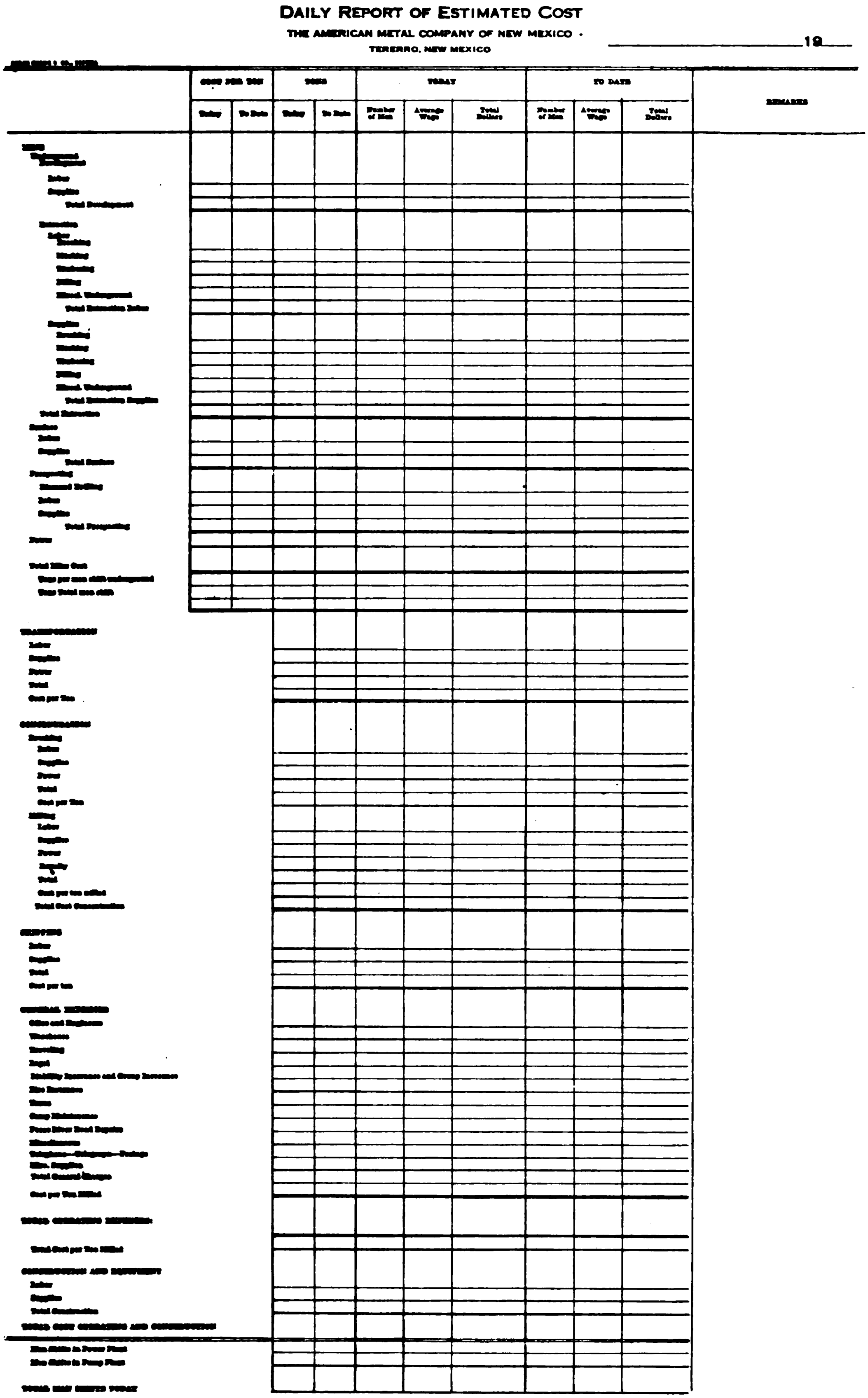
The mine is still a relatively shallow one, and several raises lead to the surface for use as fire exits. The main shaft and principal mine openings are usually wet enough to prevent fire. However, a 6-inch water line has been installed at the collar of the shaft and shaft timbers can be flooded in case of fire. Water under pressure is available in all workings underground. Hose houses have been built at strategic points on the surface so that two or more hose lines will reach any point in the plant.

Pumpmen and cagers act as fireguards during the graveyard shift. Gas masks are kept near the collar of the shaft.

Recently several men of the underground force received mine rescue training from the crew of a Bureau of Mines car, and practically all the men on the payroll took the course in first aid. The company hospital is close to the mine, and a physician can always be on hand in a very few minutes. If necessary, trained mine rescue crews can be summoned from nearby coal mines.

The chief accident hazards are the result of the loose ore and walls, which require close watching. Safety work is in charge of the mine foreman and underground bosses. A monthly bonus is paid to each boss having no lost-time accidents.

A summary of lost-time accidents for the year 1929 follows. The shift on which the accident occurs is not counted as lost time, but succeeding days are. Men are never returned to work on light duty, but are kept off until released by the doctor as fit for their regular work. Fatal and disabling accidents are weighted as shown in the table footnotes.



DAILY REPORT OF ESTIMATED COST

THE AMERICAN METAL COMPANY OF NEW MEXICO ·

TERERRO, NEW MEXICO

____________________19____

[illegible]

	COST PER TON		TONS		TODAY			TO DATE			REMARKS
	Today	To Date	Today	To Date	Number of Men	Average Wage	Total Dollars	Number of Men	Average Wage	Total Dollars	
MINE											
Underground											
Development											
Labor											
Supplies											
Total Development											
Extraction											
Labor											
Breaking											
Mucking											
Timbering											
Filling											
Miscl. Underground											
Total Extraction Labor											
Supplies											
Breaking											
Mucking											
Timbering											
Filling											
Miscl. Underground											
Total Extraction Supplies											
Total Extraction											
Surface											
Labor											
Supplies											
Total Surface											
Prospecting											
Diamond Drilling											
Labor											
Supplies											
Total Prospecting											
Power											
Total Mine Cost											
Tons per man shift underground											
Tons Total man shift											
TRANSPORTATION											
Labor											
Supplies											
Power											
Total											
Cost per Ton											
CONCENTRATION											
Crushing											
Labor											
Supplies											
Power											
Total											
Cost per Ton											
Milling											
Labor											
Supplies											
Power											
Royalty											
Total											
Cost per ton milled											
Total Cost Concentration											
SHIPPING											
Labor											
Supplies											
Total											
Cost per ton											
GENERAL EXPENSE											
Office and Engineers											
Warehouse											
Traveling											
Legal											
Liability Insurance and Group Insurance											
Fire Insurance											
Taxes											
Camp Maintenance											
Pecos River Road Repairs											
Miscellaneous											
Telephone—Telegraph—Postage											
Misc. Supplies											
Total General Charges											
Cost per Ton Milled											
TOTAL OPERATING EXPENSES:											
Total Cost per Ton Milled											
CONSTRUCTION AND EQUIPMENT											
Labor											
Supplies											
Total Construction											
TOTAL COST OPERATING AND CONSTRUCTION											
Man Shifts in Power Plant											
Man Shifts in Pump Plant											
TOTAL MAN SHIFTS TODAY											

Figure 13. — Daily cost report

Underground accidents during the year 1929

Fatal	1(1)
Permanent total disability	1(2)
Permanent partial disability	1(3)
Temporary - more than 14 days	26
Temporary - 7 to 14 days	17
Temporary - 1 to 7 days	107
Total number lost time accidents	153
Total men - days worked	154,445
Average number men employed	423
Total days lost	8,742
Frequency rate per 1,000 shifts worked	0.99
Severity rate (days lost per 1,000 hours worked)	7.08

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(1) 2,100 lost days allowed.
(2) 3,850 lost days allowed.
(3) 770 lost days allowed.

Causes of Accidents

1.	Fall of rock or ore from roof or wall	43
2.	Rock or ore while loading at working place or chute	17
3.	Timber or tools	30
4.	Explosives	5
5.	Haulage (mine cars or mine locomotives)	24
6.	Falling down chute, winze, manway or stope	11
7.	Run of ore from chute	2
8.	Drilling	5
9.	Nails and splinters	13
10.	Other causes	3
	Total	153

ACCOUNTING

The accounting system in use at the Pecos mine is the same in most respects as that in use at all other operating units of the American Metal Co., Ltd.

The features of this system that are of particular importance to the management are a daily departmental cost report in detail (fig. 13) and a daily estimated profit and loss statement (fig. 14). To complete these reports, after the necessary information has been received, requires only a part of one man's time each day.

The method of obtaining the information for the daily departmental cost report follows.

Labor

Daily time sheets are made out by each boss or foreman showing the daily rate, hours worked, and total earnings for the day of all individuals under his supervision. Each individual's earnings are distributed by the boss or foreman to the proper working place and the type of labor performed is also designated.

These daily time sheets are received by the timekeeper who posts the earnings to the payrolls and recapitulates the distribution of the labor in preparation for the report. A proportion of administrative and New York office labor is also included in the labor figures on the report.

Supplies

The warehouse disbursements are recorded on "requisition for supplies" tickets and must be signed by a responsible man in charge of the job to which the supplies are issued, and the working place given for distribution purposes.

Each day's tickets are priced the same day and delivered by the warehouseman to the office, where a recapitulation of the distribution is made conforming to that made of the labor.

The proper proportions of all fixed, accrued, and deferred operating charges are included in the report so that the costs shown thereon agree with the final costs published after the monthly closing of accounts. There are exceptions, of course, in instances where tonnage adjustments are found to be necessary or where some item of expense could not be anticipated.

Accuracy can not be attained with this system without the full cooperation of the supervisory force. For the benefit of the mine foreman and bosses, daily reports are posted showing tons, shifts, tons per shift, cost per ton, and grade.

The amounts from the cost report which constitute the day's expense are carried forward to the profit and loss statement. The method used in arriving at the value of each day's production for this statement will be described in a forthcoming paper on the Pecos mill.

As an aid to supervision, recording meters have been installed at the compressors, pumps, tram line, and saw mill. These records are of great value, showing at a glance whether operations have proceeded according to schedule, or have suffered delays which may need investigation.

Figure 15 is an organization chart showing lines of authority and coordination of departments.

Table 1.--Underground costs at the Pecos Mine, 1929, per ton of ore hoisted[1]

	Labor	Supervision	Compressed air drills and steel[2]	Power	Explosives	Timber	Other Supplies	Total 1929	Total 1928 199,508 tons	Total 1927 162,029 tons	Total three years 578,658 tons
Development and prospecting	.677	(3)	(3)	.014	.131	.099	--	.921	.817	.829	.859
Diamond drilling[4]	---	---	---	---	---	---	.297	.297	.087	.017	.146
Mining	1.656	.107	.141	.041	.128	.323	(5)	2.396	2.132	2.301	2.278
Transportation[6]	.492	---	---	.020	---	---	.199	.711	.615	.658	.663
Pumping	.092	---	---	.016	---	---	.070	.178	.120	.106	.138
General underground expense	.284	---	---	---	---	---	.101	.385	.230	.218	.285
Surface expense[7]	---	---	---	---	---	---	.010	.010	.016	---	.010
Total	3.201	.107	.141	.091	.259	.422	.677	4.898	4.017	4.129	4.379

The above costs include cost and installation of new equipment, as well as a warehouse charge which has been sufficient to create a reserve more than equal to the present warehouse stock.

1 - 217,121, tons of ore were hoisted in 1929
2 - Supplies only.
3 - Included under mining.
4 - Includes deep hole drilling.
5 - Included under general underground expense.
6 - Includes hoisting.
7 - Assaying only, surface expense directly applicable to mining being distributed in mining and other accounts.

Table 2.--Total operating costs, including tramway, crusher, mill, loading concentrates, and general charges

	1927	1928	1929	3 years to date
Tonnage mined	162,029	199,503	217,121	578,658
Tonnage milled	164,874	201,013	216,809	582,696[1]
Direct mining cost	$ 4.129	$ 4.017	$ 4.898	$ 4.379
Direct crushing cost	.092	.060	.059	.069
Direct tramway cost	.258	.445	.342	.354
Direct milling cost	1.499	1.440	1.212	1.372
Direct loading and shipping (per ton concentrates)	.298	.257	.189	.247
General charges (per ton milled)[2]	1.177	.813	.868	.936
Total operating cost per ton milled	$ 7.209	$ 6.835	$ 7.449	$ 7.169

1 - Includes 5,500 tons ore from development previous to 1927

2 - General charges include in addition to the ordinary general overhead, all insurance premiums, property taxes, proportion of head office expenses, legal expenses and cost of smelter representation.

Table 3.—Summary of direct stoping and development costs[1]

Direct stoping costs year 1929

	Labor				Supplies				Total cost per ton	Production 1927-28-29 tons	1927-28-29, cost per ton
	Mining	Mucking	Timber-	Filling	Mining	Timber-ing	Explo-sives	Tons			
Cut-and-fill........	$0.42	$0.44	$0.41	$0.25	$0.09	$0.26	$0.14	81,827	$2.01	270,873	$2.11
Shrinkage and fill..	.34	.12	.01	.01	.07	---	.15	7,846	.70	23,806	.93
Square-sets and fill	.62	.57	.79	.18	.12	.40	.15	98,902	2.83	194,030	2.66
Underhand pillars...	.62	.49	1.23	.26	.10	.42	.12	7,110	3.24	32,121	2.44
Totals...........	.52	.49	.62	.21	.10	.33	.14	195685	$ 2.41	520,830	$2.23

Direct development costs

	Cost per foot, year 1929				Cost per foot three years 1927-28-29			
	Footage	Labor	Supplies	Total	Footage	Labor	Supplies	Total
Drifts and crosscuts	11,740	$7.50	$ 2.97	$ 10.47	24,617	$8.08	$ 3.22	$11.30
Raises and winzes...	3,349	9.36	3.38	12.74	10,076	9.02	3.47	12.49
Shaft sinking.......	219	68.71	41.54	110.25	328	74.82	44.19	119.01
Stations and pockets	236	53.38	28.16	81.54	467	60.02	29.41	89.43
Totals...........	15,544	$9.46	$ 3.99	$ 13.45	35,488	$9.70	$ 4.07	$13.77

Deep-hole drilling 4,020 feet. Cost labor $1.26 per foot. Supplies $0.72 per foot.
Total $1.98 per foot.

Diamond drilling 23,114 feet. Total cost of contract price plus company expense was $2.99 per foot.

Diamond drilling 15,185 feet was from underground stations using air power - 60 holes.

Diamond drilling 7,916 feet was from surface, using gas engine power - 8 holes.

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1 - Direct stoping and development costs include all distributed accounts such as compressors, blacksmith, steel sharpening, steel and tools, timber framer expense, and nails, in all about 25 cents per ton.

Table 4.—Pecos mine - extraction costs in units of labor, power, and supplies, 1928 and 1929

Mining methods: Cut and fill, square-set, pillars, and shrinkage

	1929	1928
Tons hoisted from extraction	195,685 tons	169,796 tons
Labor (man-hours per ton hoisted from extraction):		
Breaking	.917	.774
Timbering	1.022	.864
Filling	.454	.362
Mucking	1.015	.865
Haulage and hoisting	.592	.643
Supervision	.071	.079
General	.631	.592
Totals	4.702	4.179
Average tons per man shift - extraction	1.70	1.91
Average tons per man shift - development	.54	.93
Average tons per man shift - extraction and development	1.41	1.65
Power and supplies:		
Explosives (pounds per ton 25 per cent and 40 per cent strength gelatin dynamite)	.73	.85
Timber - linear feet per ton (5 to 16 inch poles, average 8 inch diameter)	3.9	2.8
Total power - in kw. h. per ton extracted	11.779	8.099
Air compressor	6.882	5.430
Hoisting	1.302	.977
Pumping	2.727	1.211
Tramming	.779	.393
Timber framing	.089	.088
Other supplies in per cent of total supplies and power	36.54	36.07
Labor per cent of total cost	74.73	73.83
Supplies and power per cent of total cost	25.27	26.17

Table 5.--Pecos mine - development and prospecting costs in units of labor, power, and supplies, 1928 and 1929

	1929	1928
Labor (man-hours per foot)		
Sinking	68.7	67.1
Drifts and crosscuts	13.0	13.2
Stations and pockets	53.4	77.0
Raises	16.3	15.7
Total direct development	16.4	15.3
Haulage and hoisting	3.7	4.3
Pumping	2.3	1.8
Total all labor	22.4	21.4
Power and supplies		
Explosives (pounds per foot 25 and 40 per cent strength gelatin dynamite)	9.4	9.8
Timber - linear feet per foot (drifts, X-cuts and raises only)	16.4	9.8
Total power (kw. h. per foot)	46.331	39.181
Air compression	28.881	27.101
Hoisting	14.608	10.150
Haulage	2.306	1.302
Timber framing	.536	.628
Other supplies in per cent of total supplies and power	45.59	45.14
Labor, per cent of total cost	65.38	69.84
Supplies and power, per cent of total cost	34.62	30.16

The American Metal Company of New Mexico
Supplement to Daily Report of Estimated Cost

....19....

	This Month		Last Month	
	Today	To-Date	Same Date	To-Date
Tons Milled				
Tons Concentrates Produced				
Lead				
Zinc				
Value				
Zinc				
Lead				
Total Value				
Less Total Operating Expenses				
Estimated Operating Profit				
Construction and Equipment				
Estimated Net Operating Profit				
Gross Value of Heads				
Recovery				
Zinc				
Lead				
Ratio				

Figure 14.—Daily estimated profit and loss statement

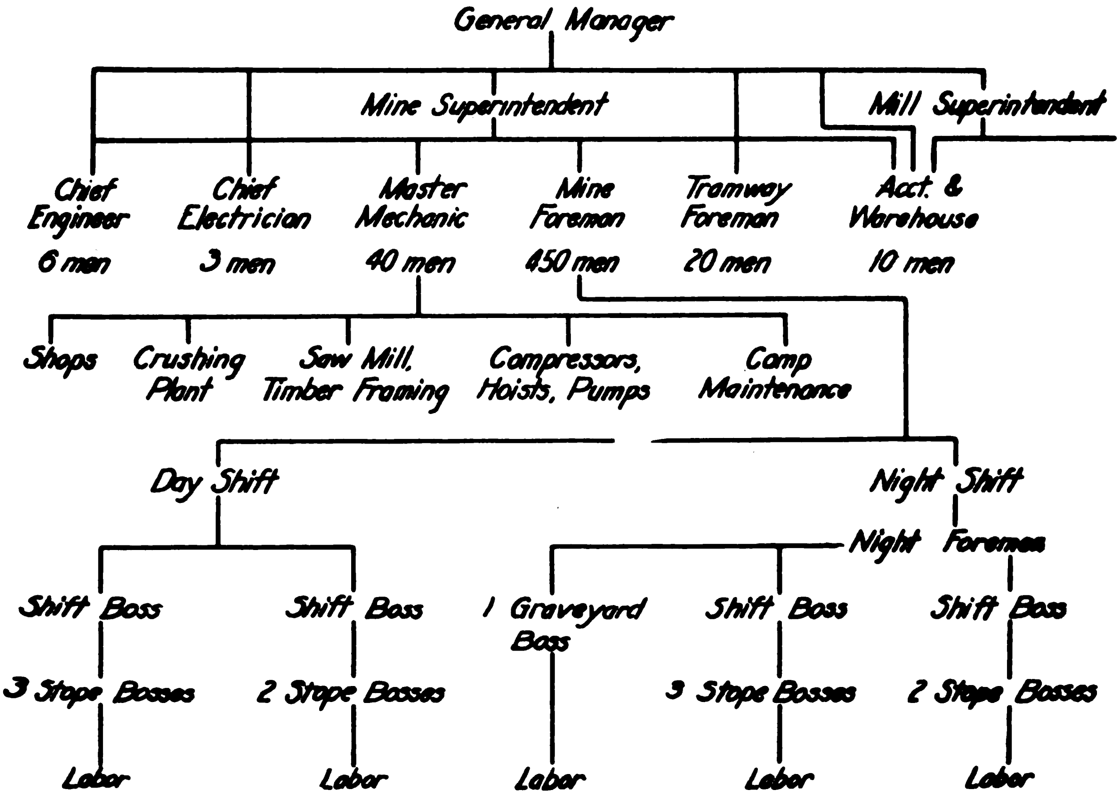

Figure 15.—Organization chart